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Photo by Bert Huntoon, Bellingham, Wash.

MT. SHUKSAN AS SEEN FROM SKYLINE EXTENSION, MT. BAKER NATIONAL FOREST HIGHWAY

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G. P. St. CLAIR, *Editor*

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SOME STUDIES OF DRILLING AND BLASTING IN HIGHWAY GRADING

By ANDREW P. ANDERSON, Highway Engineer, Division of Management, U. S. Bureau of Public Roads

POOR blasting is the rule rather than the exception in much of our highway grading work. This conclusion is based on a study of 71 power-shovel grading jobs on which solid materials requiring drilling and blasting were encountered in considerable quantities. On 43 of these jobs the blasting was decidedly deficient. On 13 of the jobs the material was fairly well broken, while on only 15, or slightly more than one-fifth, was the material so well broken as to permit rapid and fairly continuous shovel operation.

Since poor blasting is so common and materials which must be blasted before they can readily be handled by the power shovel are so frequently encountered in present-day highway grading work, the accumulated field records of the production studies made by the Bureau of Public Roads during the past six years have been reviewed in order to obtain as much definite data as possible in regard to the following points:

1. How and to what extent does the blasting affect the rate of shovel production?
2. What are the chief obstacles to adequate blasting—that is, such blasting as will permit a high rate of shovel production?
3. How and to what extent can these difficulties be overcome?

The rate of power-shovel operation depends on three factors, the average length of the dipper cycle, the amount of material handled or moved per dipper load, and the regularity or continuity of operation. Poor blasting usually means large rocks and also, very frequently, tight or even some unbroken ground. Large rocks can only be handled with much difficulty and at a very slow rate. Still further delays are often imposed on the shovel while the larger rocks are being "bulldozed" or while the unbroken ground is being reblasted. Figures 1, 2, and 3 illustrate the difficulties imposed on the shovel by poor blasting.

On the other hand, where the blasting has been so thorough that the largest dimension of the larger fragments or particles does not exceed but about one-half of the smallest inside dimension of the dipper, the rate of shovel operation can be practically the same as for operation in good common earth excavation. Figure 4 shows a cut in which adequate blasting made the work of the shovel easy. The average amount of pay material moved per dipper load, however, will nearly always be less when working in rock than when working in good common earth excavation because of the fact that rock generally takes on more swell from blasting and from being picked up by the dipper than does ordinary earth picked up by the dipper from its natural place in the cut.

Even with the most perfect blasting the rate of production in pay yardage can rarely, if ever, be expected to be as high as can be achieved in good common earth under similar conditions and with equal effort. The same rate of shovel operation can be attained, and, under favorable conditions, the same apparent or loose-volume yardage; but the actual or pay yardage measured in place in the cut will probably always be less for the rock than for good earth. Very sticky materials or earth mixed with many roots or bowlders may give much lower production than well-blasted rock, as to both size of dipper load and actual yardage per hour of operation.

RATE OF PRODUCTION REDUCED MORE THAN 50 PER CENT BY POOR BLASTING

As the thoroughness of the blasting decreases, the rate of production decreases rather rapidly. For all the jobs studied during the past three years the average operating cycle of 1 to 1¼-yard shovels working in materials classed as well to fairly well blasted rock was 23.5 seconds, while for the same size and type of shovel operating in poorly blasted materials the average shovel cycle was 33.5 seconds—an increase of over 42 per cent. Even worse conditions are frequently found, as, for example, on some very poorly blasted jobs not included in the above averages on which the average shovel cycle was 45 seconds—an increase of more than 85 per cent over well-blasted materials.

FIGURE 1.—TRYING TO MAKE THE SHOVEL TAKE THE PLACE OF DRILLING AND BLASTING IS A SURE WAY TO SHORTEN THE LIFE OF THE SHOVEL, POSTPONE THE COMPLETION OF THE JOB, AND REDUCE THE POSSIBLE PROFITS

The effect of poor blasting on the average quantity of pay material moved per dipper load is almost equally striking. For the 1 to 1¼ yard shovels the dipper load averaged 0.70 cubic yard of pay material in well-blasted rock but only 0.50 cubic yard in poorly blasted materials, a reduction of over 28 per cent. On a few jobs not included in the above averages, because of exceptionally poor blasting, the output was only about 0.4 of a cubic yard of pay material per dipper load for a 1⅛-cubic-yard dipper, a reduction of over 40 per cent. The average reduction in the rate of production for 1 to 1¼ yard shovels, not including exceptionally bad jobs, was therefore approximately 50 per cent, simply because of the decreased dipper load and the increased shovel cycle resulting from poor blasting of the material.

FIGURE 2.—A GOOD SHOVEL PLUS A GOOD OPERATOR CAN HANDLE ROCKS LIKE THIS, BUT THE COST PUTS THIS KIND OF BLASTING INTO A LUXURY CLASS WHICH NO CONTRACTOR CAN ANY LONGER AFFORD

This 50 per cent decrease in the rate of shovel production resulting from an increased shovel cycle and a decreased net dipper load does not include all the effects of poor blasting on the rate of production. More time is always lost on poorly blasted jobs than on jobs where the blasting is well done. This is brought out more fully in Table 1, which is a summary of the production studies during the past four years on 20 jobs using ⅞ to 1¼ yard shovels. Here it will be noted that the delays or individual stops due to one item alone, large rocks, were increased almost 50 per cent on the poorly blasted jobs, while the hourly production for the time the crew was on the job was reduced to less than one-half of that attained on the jobs on which the blasting was thorough.

TABLE 1.—*Average rates of production and operation on 20 rock jobs*

Condition of blasted rock for handling by shovel	Average size of shovel	Average dipper load	Shovel production per hour			Percentage of working time lost by shovel due to large rocks
			Cubic yards	Dippers		
				Number	Cycle	
	Cu. yds.	*Cu. yds.*			*Seconds*	
Good	1.02	0.68	65.5	96.3	24.4	8.7
Fair	1.08	.49	38.7	79.7	32.2	14.5
Poor	1.12	.50	31.8	63.3	36.7	16.3

COST OF SHOVEL OPERATION VIRTUALLY INDEPENDENT OF RATE OF PRODUCTION

The reduction of some 50 per cent in production simply because the material was poorly instead of thoroughly broken would not be so bad if the cost of shovel operation could be reduced in about the same proportion. This, however, is impossible. The cost of shovel operation exclusive of the hauling is practically constant so long as the crew is out on the job regardless of whether production is high or low, and with the present prevailing equipment and methods of operation even the hourly or daily cost of the hauling equipment tends to become fairly constant and only partly dependent on the rate of production. Every decrease in output, therefore, tends automatically to register as an increase in the unit production cost.

The following example illustrates the point in question. The records show two 1-yard shovels each of which had an average operating cost of about $21 per hour for the items of loading, hauling, and dump operation. At one time during a full week while one of these shovels was working in well-blasted rock at an average rate of 67 cubic yards per hour the other was working in poorly blasted rock at an average rate of only 36 cubic yards per hour. It cost the contractor, therefore, 31⅓ cents per cubic yard to load, haul, and place the well-blasted material as against a cost of 58⅓ cents per cubic yard to handle the poorly blasted material, a difference of 27 cents per cubic yard. It seems probable that if a part of this 27 cents had been expended in obtaining a somewhat closer spacing of the drill holes and the use of a slightly larger amount of explosive a considerable saving would have been effected.

In this case, which was fairly typical of average conditions, the contractor, as is usually the case, was unable to reduce either his hauling equipment or his dump crew while working in the poorly blasted material. Poor blasting almost invariably gives rise to poor hauling conditions, the main results of which are a slow hauling speed and difficult dump operation. These conditions call for proportionately more hauling equipment. Furthermore, poorly blasted material is seldom uniform. There are therefore periods when the rate of production is high enough to demand the normal supply of hauling equipment and other periods when production is so low that most of the hauling equipment is idle. The wear and tear on all the equipment is excessive when working in poorly blasted material. Time losses and delays due to mechanical troubles are also more frequent. The question, therefore, naturally arises as to how and to what extent can these losses be prevented or reduced without incurring corresponding costs.

METHODS STUDIED FOR IMPROVING QUALITY OF BLASTING

It is recognized, of course, that under the many widely varying conditions encountered in the field and with the present knowledge of the art, not all blasts will produce the much desired thorough shattering of the material, and that no rules or formulas can be given which will fit all conditions. On the other hand, it is believed that the available data are sufficiently extensive and sufficiently clear to indicate certain general guides to procedure which should be of material value, first, when work is begun on a new job or a new cut which differs materially from those in which work has already been done, and second, in interpreting and utilizing the experience gained from each successive blast. Occasionally conditions will be encountered under which it will probably be cheaper to accept a reduced shovel output, even as expensive as that is, rather than to assume the expenditure necessary to accomplish thorough breakage of all of the material; but these cases seem to be relatively few and will be discussed more fully later.

In general, the data indicate that on a large majority of grading jobs involving solid rock, hard shale, or similar materials more thorough blasting is possible,

and that such blasting will tend to decrease rather than to increase the total unit cost of moving the material.

For a proper understanding of this matter it is necessary to keep in mind the fact that the drilling and blasting involved in highway grading is very different from that encountered in any of the other common lines, such as quarrying, mining, or tunneling. The roadway and grade are laid out without much reference to such factors as the dip or strike of the strata, or to the bedding planes, seams, or stratification of the material which is to be removed. Moreover, the greater portion of the rock work is encountered in sidehill cuts so that the depth and frequently also the character of the material to be blasted varies considerably from one side of the road to the other as well as along the axis of the cut. In fact, it is not uncommon to find cuts in which no two drill holes in a row are of the same depth, and no two rows are alike. The character of the rock, as a result of weathering, dip of strata, etc., may also in extreme cases be different for practically every hole. To develop standard methods and practices which under such varying conditions will always produce the degree of breakage and shattering necessary for fast shovel operation is naturally difficult and this superabundance of variables in the conditions to which any rules or methods must be adaptable is probably the main reason for the present prevalence of poor blasting in highway grading work.

FIGURE 3.—UNDER AVERAGE HIGHWAY CONDITIONS POOR BLASTING SUCH AS THIS IS LIKELY TO INCREASE THE COST OF HANDLING MATERIAL AS MUCH AS 25 CENTS PER CUBIC YARD

LITTLE PROGRESS MADE IN IMPROVING METHODS

While notable progress has been made in effecting higher rates of production in practically all other lines of highway work, the rate at which drilling and blasting is handled has shown no appreciable progress during the past six years, for which the bureau has available definite records based on production studies on a large number of going projects. These data, while not sufficiently extensive to be entirely conclusive, are nevertheless sufficient to indicate that for identical equipment there has been during this period no material increase in the efficiency with which the drills are operated, in the footage of holes drilled per hour of actual operation, or in the rate of production of the shovel while handling blasted material. Nor is there any substantial evidence of a definite decrease in the time losses in drill operation, in the amount of either drilling or explosives required per cubic yard of pay excavation, or in the time losses imposed on the shovel by the operations of drilling and blasting. However, the trend toward the use of larger shovels has been rather marked and the proportionate loss of time has therefore been somewhat reduced.

From these indications one might argue that further progress in handling rock excavation in highway grading work is impossible or at least improbable; but, while the records of past performance are far from inspiring, they are not as hopeless as these generalities would indicate. Out of the 71 jobs there were 15 which attained practically as high a rate of production in rock as is usually secured in common excavation. One group of 1 to 1¼ yard shovels working in material classified as "good to fair common" operated on an average cycle of 21 seconds while moving 93,000 cubic yards of material, while another group of shovels of the same size working in material classified as "rock, well blasted, very few pieces too large for dipper" also operated on an average cycle of 21 seconds while moving 48,000 cubic yards of material.

FIGURE 4.—ALMOST ANY ROCK, WHETHER STRATIFIED OR MASSIVE, AND NO MATTER HOW HARD OR TOUGH, CAN BE SO THOROUGHLY BLASTED THAT IT CAN BE HANDLED BY THE SHOVEL AT ABOUT THE SAME RATE AS GOOD COMMON. THIS IS THE SAME KIND OF ROCK AS IS SHOWN IN FIGURE 3

The average dipper load of the shovels working in common was 0.66 cubic yard of pay material while for those working in the well-blasted rock it was 0.61 cubic yard. Thus, for the time these shovels were actually digging, the rate of moving cubic yards of pay material in well-blasted rock was 92 per cent of the rate maintained by the shovels working in good common. But, probably because of a somewhat better supply of hauling equipment, the time losses on the rock jobs were about 6 per cent less than where good common was being handled, so that the actual production per hour on the rock jobs was only 2 per cent below that attained on the earth jobs. Based on the total time the crews were on the job, the actual production of pay yardage was 57.5 cubic yards per hour for the shovels working in good common and 56.4 cubic yards per hour for the shovels working in well-blasted rock. The ordinary time losses or delays, while rather unusually high, were very nearly the same for both sets of shovels, 49 per cent for those working in common excavation and 46 per cent for those working in rock. In both cases the major portion of these time losses was due to insufficient hauling equipment.

For these reasons it seems necessary to examine a little more closely the conditions which exist in drilling and blasting for highway grading work. In the first place, we find that on the great majority of jobs on which rock is encountered, it is usually covered with more or less overburden so that advance determination of such important factors as the exact amount, char-

acter, and condition of the rock, the dip of the strata, location of seams and bedding planes, etc., is difficult. Sometimes considerable rock is encountered where none had been expected. As a rule, the depth of rock to be removed in these cuts is comparatively small, seldom above 20 feet, on many jobs rarely over 4 or 5 feet, and frequently only a foot or two. The limitations placed on either over or under breakage on the side slopes, and frequently even in regard to the grade line, are rather numerous.

Careful consideration must also be given to preventing injury to persons, passing traffic, and neighboring improvements. The shooting must, therefore, as a rule, be as light as possible. But light shooting and thorough breakage of the material can only be achieved through correct depth and placement of drill holes which have been properly loaded with the right amount of suitable explosives and then properly fired. Even heavy shooting frequently fails to break up a hard or tough material properly unless the charges have been both placed and spaced correctly. (See fig 5.) Success

FIGURE 5.—IT TAKES MORE THAN GOOD INTENTIONS TO BREAK HARD AND TOUGH ROCK. THE BLAST JUST FIRED DID LITTLE MORE THAN BREAK THE MATERIAL INTO LARGE BLOCKS, EACH OF WHICH NOW FORMS A SEPARATE PROBLEM

can be attained only through a correct coordination of all of these factors while a substantial error in any one is almost certain to result in a costly failure. Since drilling is usually the most expensive item in the type of rock work encountered in highway grading, while improper location, spacing, or depth of the drill holes is also the most frequent cause of poorly broken material, we shall discuss this item in more detail.

METHOD SUGGESTED FOR SPACING DRILL-HOLES

Figure 6 shows a part of a rock cut where two rows of holes have been drilled. It will be observed that the holes are staggered. The relations between the spacing of holes, depth of cut, and depth of drill hole below grade are expressed as follows:

a = Ratio of depth of cut to spacing of holes across cut.
b = Ratio of depth of cut to spacing of holes from face, or of rows across cut.
c = Coefficient of depth of drill hole below grade. Its value, for most materials, lies between one-fourth and one-half, except in very shallow holes.
d = Depth of cut at drill hole.
ad = Distance between holes across cut.
bd = Spacing of rows or distance of holes from face of cut.
cad = Depth of drill hole below grade to which rock is to be removed.
$d+cad$ = Total depth of drill hole.

In this arrangement the drill-hole spacings parallel and perpendicular to the face of the cut are not equal, but each is a function of the depth. The depth drilled below grade is a function of the spacing rather than the depth.

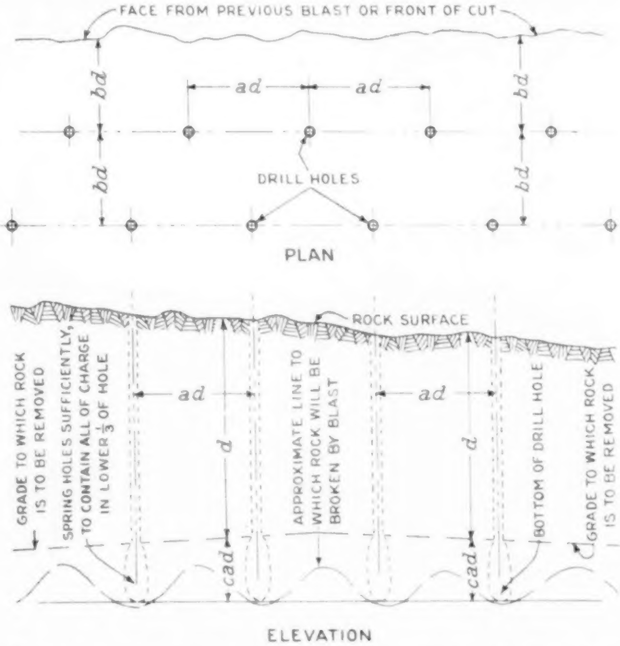

FIGURE 6.—SUGGESTIONS FOR PLACING DRILL HOLES. DIAGRAM SHOWS PART PLAN AND PART ELEVATION OF NEARLY LEVEL THROUGH CUT

A review of all the field records for jobs working in reasonably homogeneous material shows that the spacing between the holes in a row across the cut averaged about three-quarters of the total depth and usually varied from one-half to slightly more than the depth of the holes. The spacing of the holes from the face or between rows averaged slightly less than the spacing between the holes in a row. In such materials as ordinary shale, disintegrated granite, and hardpan the spacing in deep cuts sometimes equaled twice the depth of cut but was usually about one to one and one-half times the depth of cut. However, there was a rather strong tendency to adopt and maintain a fairly constant spacing for any given cut regardless of the variation in the depth of individual holes.

Too frequently the exact placement of a hole was determined by the existence of a good place to drill rather than by the relative distances to adjoining holes. The depth drilled below grade was most frequently an arbitrary constant for any given cut with but little or no variation for different depths of the individual holes. Thus, on one job on which the depth of cut varied from 2 to 10 feet and the spacing from 2½ to 6 feet, the depth below grade was always 2 feet. On a second job on which the depth of cut varied from 5 to 20 feet and the spacing from 4 to 10 feet, the depth below grade was always 2 feet. On a third job in disintegrated granite with cuts from 5 to 24 feet and the spacing about equal the depth, the depth below grade was always 3 feet, while on a fourth job with cuts mostly 10 to 20 feet deep and a spacing of about 12 feet between holes and 8 feet between rows, the depth below grade was always 4 feet. A further consideration of Figure 6 will help to illustrate why such practices are likely to result in improper fragmentation or shattering of the material, and high unit costs.

UNIT COST HIGH FOR SHALLOW CUTS

If the distances between holes and rows as well as the depth of the holes are given in feet with the designations as indicated in Figure 6, we note that the volume of material above grade line may be expressed in cubic yards for each hole as:

$$V=\frac{abd^3}{27};$$

while if D be used to designate the linear feet of drilling required per cubic yard of pay material, we have:

$$D=\frac{27\ (1+ac)}{abd^2}$$

Thus, within those limits in which the ratio of the spacing of holes and rows as well as depth below grade holds true, the volume of pay material per drill hole varies directly as the cube of the depth of the cut and the amount of drilling per cubic yard of pay material varies inversely as the square of the depth of the cut. The amount of explosives required per cubic yard of any given material, however, varies only slightly with the depth of the hole. These facts help to explain why the unit cost of removing shallow cuts of hard rock is so high.

If we consider, for example, four cuts of the same kind and character of uniform solid rock, the first of which is 2 feet deep, the second 4 feet, the third 8 feet, and the fourth 16 feet deep, in which both rows and holes can be spaced at three-quarters of the depth of the cut and the depth to which the holes must penetrate below the grade is one-fourth of the spacing between holes, then the amount of drilling, in linear feet, required per cubic yard of pay material for each cut is as follows:

2-foot cut	14. 25
4-foot cut	3. 56
8-foot cut	. 89
16-foot cut	. 22

Thus, if the cost of drilling be 30 cents per linear foot, as is frequently the case, then the cost of drilling alone is \$1.07 per cubic yard of pay material for the 4-foot cut, \$0.27 for the 8-foot cut, \$0.07 for the 16-foot cut, and the entirely prohibitive figure of \$4.27½ for the 2-foot cut. Even in the case of the 4-foot cut, the cost of the drilling alone is higher than the price frequently bid for moving solid rock, while on the other hand in the case of the 16-foot cut the unit drilling cost becomes but a relatively small item in the total cost of moving the material.

PROCEDURE MUST BE VARIED TO FIT ACTUAL CONDITIONS

The amount of explosives required per cubic yard of material tends to remain much more nearly constant, increasing only slightly as the depth of the holes decreases. Both theory and practical experience indicate that in shallow cuts it is cheaper to drill proportionately much deeper below grade than is advisable in the deeper cuts. This, of course, requires more explosives in moving nonpay material but decreases very materially the unit cost per cubic yard for the total combined operations of drilling and blasting. Experience has also shown that in very deep cuts of tough, very blocky, or nonuniform rock thorough breakage can not readily be accomplished if the holes are spaced as far apart as theory would indicate to be possible. While in most rockwork encountered in highway grading the spacing both between holes in transverse rows and of rows from the face or from each other bears some fairly definite relation to the depth of cut in that particular material, there is both an upper and a lower limit fixed largely by considerations of cost and the difficulty of effecting the proper degree of fragmentation. These limitations can be expressed as follows.

The spacing of drill holes both from the face and from each other should be proportioned to the depth of cut at that point, except that only in the most exceptional cases where is it impractical to drill the holes at least 4 feet deep need the spacing of rows ever be less than 3 feet from the face or between rows. For most rock the spacing between holes need never be less than 4 feet, provided that the total depth of the hole is not less than 4 feet and penetrates at least a foot and a half below the level to which the material must be removed. In very tough or blocky material, a spacing of over 10 to 12 feet will usually require the drilling of secondary holes to about one-third to one-half the depth of the cut in the middle of the areas formed by a hole at each corner. These secondary holes, as well as any supplementary charges in the stem of the main holes, are then fired simultaneously with the main charges. This method should give good breakage for cuts up to 20 feet, which probably is close to the limit of the present economic depth of a single lift in most rock materials. In sidehill work considerable care must be taken in arranging the layout of the drill holes in order to prevent a too wide spacing for the shallow holes.

As has already been indicated, it is usually found advisable to drill the very shallow cuts considerably below grade in order to permit a wider spacing and so reduce the drilling cost. While a strict adherence to the rather general practice of spacing the holes three-quarters of the depth of the cut would, in a 2-foot cut, require over 14 feet of hole per cubic yard of pay material, according to the formula given above, the drilling of 4-foot holes 4 feet apart and 3 feet from the face would reduce the drilling to about 4.5 feet of hole per cubic yard of pay material. The total amount of powder required per cubic yard of pay material would probably be about doubled, but this would be largely compensated for by the saving in blasting caps, wiring, and time and trouble of loading. Where hard shooting is permissible the total cost of drilling and blasting a cut as shallow as 2 feet can be still further reduced by additional increases of the depth and spacing of the holes and the amount of explosives. This is especially true for materials which are not very difficult to fragment or shatter properly. Increasing the depth of the holes in the above 2-foot cut to 5 feet and the spacing to 5 and 4 feet, respectively, would require only about 3.38 feet of drilling per cubic yard of pay material, a saving of about 1.1 feet of drilling at the cost of an increase of about one-fourth to one-half pound of explosives per cubic yard of pay material. Figure 7 illustrates the difficulties of drilling and blasting in shallow cuts of hard rock.

On a few jobs it was found advantageous to use two different depths of hole. For example, for one blast the first holes were drilled about 18 feet deep (3 feet below grade) with a spacing of 12 feet across the cut and about 11 feet between the rows which were not staggered. A second set of holes was then drilled in the center of the squares formed by the four holes to a depth of about 7 feet, or not quite through the hard top strata which rested on a thin seam of softer material.

The deep holes were sprung and the principal charge placed at the bottom with a smaller charge in the stem above the soft material. The shallow holes were only lightly sprung so that the charge occupied slightly more than one-half the depth. The explosive was 40 per cent dynamite and all charges were fired simultaneously.

FIGURE 7.—SHALLOW CUTS IN HARD ROCK ALWAYS PROVE EXPENSIVE. ALTHOUGH THE CONTRACTOR SPENT $2 PER CUBIC YARD ON DRILLING AND BLASTING IN THIS CUT (MOSTLY 2 TO 4 FEET DEEP), THE ROCK WAS STILL SO POORLY BROKEN THAT THE COST OF GETTING IT INTO THE FILL ADDED OVER ANOTHER DOLLAR

DRILLING COSTS AND RATE OF PRODUCTION DISCUSSED

On typical projects the average cost of drilling in fairly hard uniform rock with the customary jackhammers (see fig. 8) was nearly 30 cents per linear foot of finished hole. Largely because of the varying nature of the rock the drilling cost per foot varied over a wide range and occasionally reached about double this figure. One rather extensive job on which the average depth of the holes was 5 feet with most of the cuts from 2 to 5 feet deep, required 2.85 linear feet of drill hole per cubic yard of rock at a cost of $1.60 per cubic yard or 56 cents per linear foot of drill hole for the drilling alone. On the other hand, on some jobs involving mostly deep holes the drilling per cubic yard was sometimes as low as 0.1 foot. There is no definite indication that the net hourly rate of drilling is appreciably faster for shallow than for deeper holes, except in materials which give trouble from sticking or broken drills. The slightly larger time losses in the deeper holes from changing steel, blowing holes, etc., seem to be largely compensated for by the fewer moves until a depth of about 20 feet is reached, after which the rate slows up. All but five of the rock jobs studied used air-driven jackhammer drills weighing about 70 pounds each and all statements and conclusions are based on the use of drills of approximately this size and type.

The rate of drilling varies with the material, the type of equipment, and the management. On some individual jobs over 60 per cent of the time the drilling crew was out on the job was unproductive. The rate of production per elapsed hour was therefore low on these jobs, regardless of the kind of material. Table 2 shows the average rates obtained on all the jobs for which complete data were available, the net rate while the drill was actually in operation, and the causes of the more persistent delays.

The materials listed in Table 2 have been grouped in accordance with their apparent ease or difficulty of drilling as indicated by these studies rather than according to the usual classification. The table shows that while the average actual cutting rate of the drills in some materials was nearly 75 per cent more than in others, the actual production of linear feet of completed drill hole per hour during the time the crew was out on the job was much more nearly uniform and the maximum difference in actual production of finished drill hole per drill was less than 35 per cent.

TABLE 2.—*Average rates of production and time losses for drilling in various materials: 60 to 70 pound jackhammers used in most cases*

Item	Sandstone, hard shale, hardpan, disintegrated granite	Granite, porphyry, felsite, trap, basalt	Hard limestone, stratified but of uniform hardness	Limestone, stratified, with clay seams	Weighted general average of studies
	Feet	*Feet*	*Feet*	*Feet*	*Feet*
Feet drilled per hour, total study time	13.8	14.8	15.5	11.5	14.2
While drill was in operation	36.2	28.9	24.2	21.2	26.1
DELAYS DURING STOP-WATCH STUDIES [1]					
Change drills	12.1	9.8	7.1	7.8	8.9
Clean or blow holes	23.4	10.0	4.0	14.0	11.1
Steel stuck or broken	15.8	7.6	5.3	5.3	7.3
Move to new hole	2.5	4.9	7.2	3.4	4.0
Springing holes		1.5	1.0	1.7	1.4
Operator	3.2	4.4	4.7	4.0	4.0
Mechanical trouble	.7	2.2	3.2	2.0	2.0
No steel	.2	3.8	1.7	.5	2.6
Miscellaneous	4.1	4.4	1.9	7.1	4.3
Total lost time while out on job	62.0	48.6	36.1	45.8	45.6

[1] Delays are tabulated in percentage of available working time

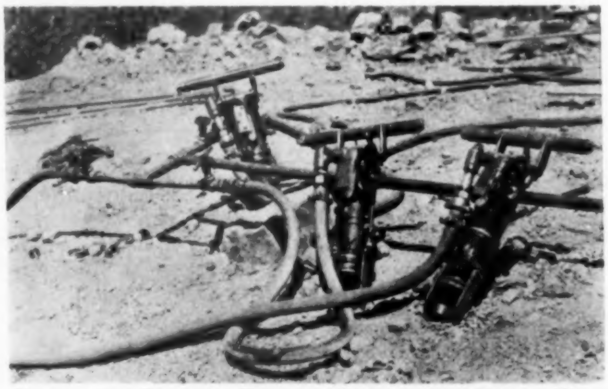

FIGURE 8.—GENERAL TYPE OF JACKHAMMER DRILLS USED IN HIGHWAY WORK

The time losses or delays in the easily drilled rocks of the first group were so large as more than to counterbalance the gainful effects of easy drilling. The chief delays were due to cleaning the holes and stuck or broken drills. These causes accounted for about two-thirds of the total time losses in the soft or decomposed materials, as against one-third for the same items in the igneous rocks of the second group. The poor showing in regard to actual cutting rate in the stratified limestones was largely due to the caution continually exercised by the drill operators to avoid crooked holes and stuck drills. That such caution can be made to pay is evident from the fact that while the cutting rate in hard stratified limestone was 25 per cent less than for the rocks of the first group, the actual production in finished drill holes was over 12 per cent greater. The generally rougher country in which the igneous rocks are usually encountered is reflected in the difficulty of keeping the drill operators supplied with steel. For the igneous group 3.8 per cent of the otherwise available

working time was lost on this account, as against a maximum of 1.7 per cent for any other group.

AIR PRESSURE AND QUALITY OF DRILL STEEL IMPORTANT

There are apparently several other conditions which influence the rate of drilling but upon which definite data are difficult to obtain. Thus, the rate at which the drill will cut in a given rock depends on the air pressure at the drill, the type and condition of the drill bit, and the hardness of the cutting edge or the length of time that the cutting edge remains fit for rapid cutting.

High air pressure is not of itself a guarantee of a high drilling rate for the average job. In hard rock a working pressure of over 100 pounds per square inch at the drill is very likely to cause excessive drill breakage. In a soft, seamy, or highly stratified rock with a large angle of dip, working on full open throttle at this pressure is likely to cause excessive sticking. In rock of this kind a working pressure above 100 pounds per square inch at the drill is not necessary. For very hard rock the present knowledge and facilities for obtaining the proper degree of hardness and toughness in the sharpened steel to withstand this pressure are not as yet available to the average highway job.

More attention should therefore be given to perfecting both the steel now used for drills and the methods of heat control and manipulation employed in the drill-sharpening outfits commonly found in highway work. Not only is there great need for a drill which will permit advantage to be taken of the higher cutting rate in hard rock which can be achieved by the use of higher air pressure, but there is also need of a more adequate drill steel or a means of hardening and tempering present steels so that they will better maintain their cutting edges and proper gage sizes.

Table 3 shows the effect of air pressure at drill on rate of cutting hard granite and hard stratified limestone. The stratifications, however, were all practically horizontal and therefore did not add to the difficulty of drilling.

TABLE 3.—*Effect of air pressure at drill on rate of drilling with 70-pound jackhammer in good condition. Holes 8 to 15 feet in both studies*

HARD LIMESTONE WITH HORIZONTAL STRATA

Working pressure at drill	Rate of drilling while drill was in actual operation	Number of 1-hour studies
Pounds per square inch	*Feet per hour*	
56	11.0	1
60 to 70	23.8	22
70 to 80	22.8	44
Above 80	28.9	2

HARD GRANITE

Working pressure at drill	Rate of drilling while drill was in actual operation	Number of 1-hour studies
45	1.5	1
50	8.7	3
60 to 70	13.9	6
75 to 87	21.8	5

On a great majority of the jobs studied the working air pressure at the compressor ranged from 80 to 100 pounds per square inch. Thus, only one job was found which used a regular working pressure of 125 pounds and only one job using a regular working pressure below 70 pounds per square inch. But the pressure at the compressor may be quite different from that at the drills. In general, too little attention is given to the size or condition of the pipe and hose lines. Air leaks were numerous in most lines. Gage readings on one job showed an average loss of 1 pound of pressure for each 16 feet of ¾-inch hose or ¾-inch ordinary iron pipe line when supplying one drill at about 75 pounds pressure. However, but few jobs were found with a pipe line as small as this. The more common size was 1 inch for the pipe, although the use of ¾-inch hose was fairly common. A few jobs on which four to six drills were in regular use from a common pipe line used 2-inch pipe, while one job had a 2½-inch line. Pipe lines as long as 1,200 feet were found in use. Needless to say, pipes of this length must have very tight unions and be of sufficient size for the number of drills to be supplied, or the working pressure will be very low.

TIME LOSS IN UNPRODUCTIVE OPERATIONS DEPENDS ON OPERATING SKILL, EQUIPMENT, AND CHARACTER OF ROCK

On present jobs from 7 to 12 per cent of the working time is lost in changing drills, although it is known that the practice of using each steel until its full depth has been reached is far more common than that of changing the steel when it has become dulled to such extent that the rate of penetration is materially reduced. Thus, the common practice is to cut the steel into lengths which are multiples of 1½ or 2 feet, the latter being the most common. But hard and difficult rock will often dull the steel so much in drilling 1 foot that the time required for the following 6 inches is equal to that required for drilling the first foot. About the only times regular changes are made before the drill length has been attained is when the drills regularly lose so much gage that the following drill will not work readily. And even these conditions are usually blamed on the blacksmith, who is importuned to increase the die.

The studies show that under normal field conditions the drill is cutting rock only from 35 to 60 per cent of the total time the operator is on the job. The rest of the time is largely consumed in performing a number of necessary but unproductive operations, such as changing steel, blowing the hole, moving to the next hole, oiling the drill, etc. Frequently additional operations become necessary or are added to the duties of the drill operator, such as removing a stuck steel, clearing overburden, bringing up steel, or springing the drilled holes. (See Fig. 9.) The amount of time consumed in performing each of these operations varies with a number of factors, among which the skill and experience of the operator and the kind, character, and nature of the rock play a very important part. The time studies show that some operators regularly consume at least twice as much time in changing steel as do some others, while in certain kinds of rock the unskilled or inexperienced operator frequently loses several hundred per cent more time in extricating stuck steel than his more skilled and experienced coworker. The amount of time lost in performing the various necessary unproductive operations is also dependent on the character of the rock. In rock in which the drill will cut at the rate of a foot in two minutes, the number of steel changes per hour will ordinarily be about twice as many as in a rock in which the drill will only cut at the rate of a foot in four minutes. On the other hand, in soft rock more time is usually required for such operations as blowing or cleaning the hole and from steel getting stuck, while in very hard rock much time is frequently lost from broken steel, especially if the air pressure at the drills is too high.

The actual time in which each individual operation can be performed also varies with several factors. For example, aside from the skill of the operator, the time required to exchange steel varies with the depth of the hole. The following is a typical set of stop-watch readings of the exchange time for an excellent operator while drilling a hole 15 feet deep in fairly hard but uniform granite: 20, 22, 25, 32, 40, 40, 47, giving an average exchange time of 32 seconds for the hole. The first and second exchanges of steel are frequently made

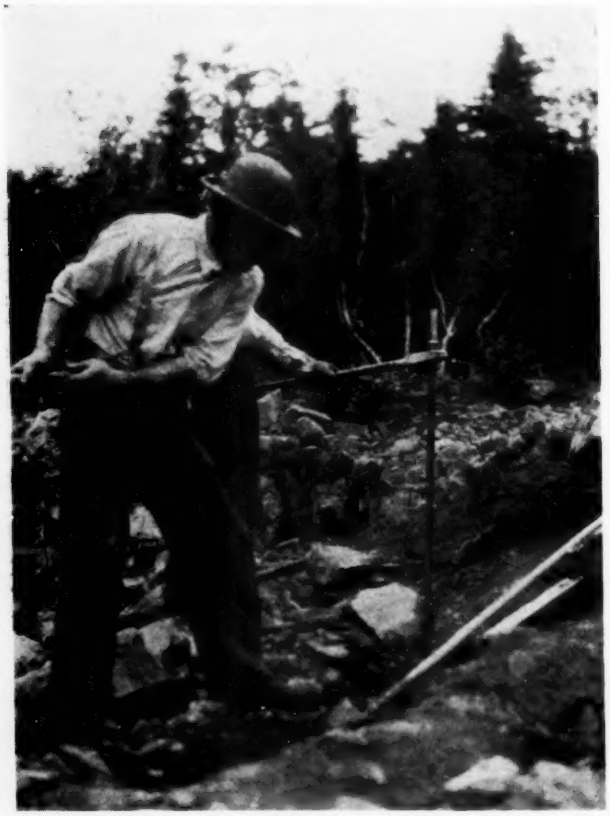

FIGURE 9.—IN SOME ROCK AN INEXPERIENCED OR CARELESS OPERATOR WILL SPEND MUCH OF HIS TIME REMOVING STUCK STEEL. POOR SHARPENING EQUIPMENT OR A CARELESS BLACKSMITH WILL INTENSIFY THIS TROUBLE

in 15 to 20 seconds and single individual readings as low as 12 seconds are on the records. The increase with depth is rapid, and a 20 to 25 foot steel is seldom exchanged in less than a minute with the average time nearer 2 minutes. If the steel happens to stick or bind in the chuck, as is frequently the case, the exchange time for that particular change may readily be doubled or even trebled. This is the reason why the records of many thousand stop-watch studies show an average exchange time of about 45 seconds for holes from 12 to 15 feet in depth, while the individual readings range from 12 seconds to somewhat over 6 minutes, and the percentage of the operator's time consumed in exchanging steel varies from less than 3 per cent to over 25 per cent with an average of 8.9 per cent for 20 fairly large jobs. (See Tables 2 and 3.)

The contractor who has much rockwork should therefore find it profitable to give considerable attention to the training of the drill operators to perform all necessary operations in the shortest possible time. The drilling crew is really only productive when the drill is actually cutting rock. The various operations which interfere with actual production should therefore be reduced to the lowest possible limit. The contractor's interest must not end with simply insuring a high utilization of the available time. He must also see that the utilized time is converted into the greatest possible number of linear feet of properly placed and spaced holes. This requires the constant maintenance of adequate air pressure at the drills, together with an abundant supply of properly sharpened and correctly tempered steel of the right lengths within easy reach of the operator. One of the surest ways to make certain a low production from the drilling crew is to permit any one of the following to occur: Low air pressure at drills, too soft or too hard steel, poorly sharpened or poorly gaged steel, or requiring the drill operator to bring up his own steel or to move a heavy overburden in order to locate the holes. Under many conditions any one of these can readily reduce production more than 25 per cent, and if two or more are allowed to occur simultaneously production may readily drop to one-half of what would otherwise be possible.

AMOUNT OF EXPLOSIVES USED VARIES WIDELY

The amount of explosives required per cubic yard of identical pay materials is generally larger in highway grading than in most lines of rock work, since it is so essential that all the material be broken into fragments small enough to permit fast and steady operation of the shovel. In most highway work this does not mean the use of exceptionally large single charges but rather closer spacing of holes, lifts of moderate depths, and a careful proportioning and placing of the charges of the proper kind of explosive so as best to utilize the full force of the explosion in loosening and shattering the material. In very hard, tough material the holes should always be sprung so as to form a cavity or chamber of sufficient size to accommodate most of the charge at the bottom of the drill hole. The same is also true of deep holes in the softer rocks, except when the danger of losing the hole by the springing is too great. To form this cavity in hard rock may require the firing in each hole of as many as five or six gradually increasing successive charges of high-strength explosive. On most of the jobs studied one to three successive charges usually proved sufficient to spring the holes, although one single hole has a record of 16 successive charges.

Table 4 shows for a number of jobs the amount of explosives used both for springing and blasting and also certain other pertinent data as to the type of explosive, average depth of holes, and the condition of the material after the blast. The two columns showing respectively average depth and average spacing of the holes give the arithmetical averages for all holes for which complete data were available. Since sidehill cuts predominated on all but three jobs (1, 2, and 5) these figures give only a roughly approximate measure of either the actual depth or the actual spacing of the individual holes and were added only to give a general idea of the nature of the work.

Table 4 shows that the average amount of explosive used per cubic yard of rock on each of these jobs varied from 0.50 to 1.90 pounds, and that the job using the smallest amount of explosive resulted in good breakage of the material while the breakage on the job using the most explosive was very poor. This helps to emphasize the fact that good breakage or fragmentation can be obtained fairly easily in some materials, but is difficult to obtain at all in others, and especially so in shallow

cuts of tough material. The poor results on job No. 10 can be ascribed very largely to the fact that the holes were not sprung. Consequently, the charge in a 5-foot hole usually came within about 18 inches of the top, which was insufficient space for proper tamping even when 60 per cent dynamite was used. Near the close of the studies, springing the holes three times with one-half, one, and two 1½ by 8 inch sticks of 60 per cent dynamite was tried, with the net result that the use of the same total amount of explosive per cubic yard gave a better fragmentation with a spacing of 4 by 4½ feet, than had been obtained from a 3 by 3 foot spacing of unsprung holes. This wider spacing required only half the amount of drilling previously necessary.

Table 4.—*Average amount of explosives used per cubic yard of pay material on 17 typical jobs*

Job No.	Explosive per cubic yard: Dynamite or gelatin: Springing holes	Explosive per cubic yard: Dynamite or gelatin: Blasting	Explosive per cubic yard: Black powder: Blasting	Explosive per cubic yard: Total	Condition of material for moving with shovel	Kind of material	Average depth of holes (approximate)	Average spacing of holes (approximate)
	Pounds	*Lbs.*	*Lbs.*	*Lbs.*			*Feet*	*Feet*
1	0.04	......	0.85	0.89	Poor	Soft and seamy granite.	30	25 by 18.
2	.55	......	.98	1.53	Poor	Hard basalt	18	19 by 19.
3	.27	0.44	......	.71	Good	Massive granite	14	10 by 8.
4	.01	......	.55	.56	Fair	Conglomerate	15	Variable
5	.25	.85	......	1.10	Poor	Stratified limestone	16	15 by 12.
6	.06	......	1.06	1.12	Good	Shale	13	12 by 12.
7	.02	.03	.45	.50	Good	Disintegrated granite.	10	Variable.
8	.15	.16	.67	.98	Fair	Stratified limestone and shale.	12	10 by 10.
9	.02	.17	.50	.69	Good	Seamy basalt	21	15 by 15
10	(1)	1.90	......	1.90	Very poor	Hard granite	5	3 by 3.
11	.25	.75	......	1.00	Poor	Very hard ganite	14	8 by 8.
12	[2] .55	1.00	......	1.55	Good	Very hard felsite	13	7 by 7.
13	.14	.59	......	.73	Fair	Hard schist	12	9 by 9.
14	.10	.50	......	.60	Good	Foliated granite	10	10 by 10.
15	.18	.75	......	.93	Fair	Hard granite	16	8 by 8.
16	.15	.87	......	1.02	Good	Very hard granite	16	8 by 8.
17	.10	.47	......	.57	Good	Soft granite	18	10 by 10.

[1] Near close of studies springing was begun using approximately 0.30 pound of 60 per cent dynamite per cubic yard for springing and 1.60 pounds for blasting.
[2] Part of this was used in burning the deeper holes to keep drills from sticking.

CHARGES SUGGESTED FOR FIRST TRIAL SHOTS

In shale, hardpan, and some soft or partly disintegrated rocks the use of a black powder is usually preferable, as these blasting powders are much slower in action and give a very large volume of gas which exerts a lifting or heaving action that is very effective in this type of material. The ordinary black powder can only be used in dry holes; and, since both sidehill and through cuts in road work are likely to be more or less wet, a low-strength, slow dynamite or blasting gelatin must generally be used. Figure 10 shows the results produced by an explosive charge of this type. Because of the prime importance of insuring good fragmentation, the use of a rather high-strength dynamite or blasting gelation is generally found most satisfactory for the harder rocks. For the first trial purposes, a blasting charge of 1 to 1¼ pounds of 50 per cent or 60 per cent dynamite, or its equivalent, per cubic yard of burden is suggested for untried materials of a hard and tough character. These suggestions are given in outline form in Table 5.

For the first trial only enough holes should be drilled and fired to make a fair test of what this spacing and loading will do in the given material. From this evidence either the spacing or the loading or both should be modified as the results indicate to be necessary or advisable. Table 5 is intended for trial use only where the depth of cut is between 5 and 20 feet.

Table 5.—*Suggestions for spacing of drill holes and amounts of explosives for the first trial shots in unfamiliar materials where hard shooting is permissible*

Type of material	Spacing of drill holes [1]: a	Spacing of drill holes [1]: b	Coefficient [1] of depth of hole below grade, c	Pounds of explosive [2] per cubic yard of burden	Use an explosive equivalent to—
Any rock exceptionally difficult to shatter.	½	½	½	1⅛	60 per cent straight dynamite.
Most hard, dense, unweathered ledge rock.	⅔	⅔	⅓	1	50 per cent straight dynamite.
All medium hard, weathered, or partly disintegrated rock; those which shatter readily and very hard shales.	¾	⅔	⅓	1 to 1½	40 per cent straight dynamite, or blasting powder if holes are dry.
Ordinary shale, hardpan, and similar materials.	1¼	1	¼	⅞ to 1	Blasting powder if holes are dry; otherwise 20 per cent straight dynamite.

[1] See Figure 6 and text, p. 296.
[2] The loading suggested is net amount of explosives for actual blast and does not include amount required for springing holes.

Figure 10.—For Most Shales, Hard-Pans and Partly Disintegrated Rocks an Explosive Having a Slow, Heaving Action Is Best. This 18-Foot Lift Was Thoroughly Broken, Although the Only Surface Indications Were a Few Cracks and a Heave or Bulge Above the Original Ground Line

Following is the procedure for finding the actual amount of explosive which should be placed in each hole. Measure the depth of the hole, its distance from the next hole in the same row, and also the distance from the hole to the face or to the next row of holes. Multiply these three distances together and, if in feet, divide the product by 27. The quotient is the "burden" on this hole in cubic yards. Multiply this burden by the amount of explosive to be used per cubic yard. This gives the amount of explosive to be placed in that hole. Thus, if a hole is 13 feet deep and the spacing is 6 and 7 feet, respectively, then the burden is

$$\frac{13 \times 6 \times 7}{27} = 20.2 \text{ cubic yards.}$$

If the rock is very hard and tough so that 1⅛ pounds of dynamite are required per cubic yard, then the charge for this hole will be

$$20.2 \times 1\tfrac{1}{8} = 22.7 \text{ pounds.}$$

For shale, hardpan, or soft or partly disintegrated materials the trial charge might well be either a coarse blasting powder; if the holes are wet, a slow dynamite should be used. If the ground will permit springing without the loss of too many holes they should always be sprung sufficiently to chamber practically the entire charge and should be loaded and tamped as indicated in Figure 11.

When there are intervening or alternating strata of hard and soft material, closer than normal spacing will usually be required and a portion of the charge in the form of dynamite should be placed in the stem of the hole within the hard strata as indicated in Figure 11. These auxiliary charges in the stem should all be fired simultaneously with the main charge by means of independent caps and primers wired to the same circuit. This method should give fairly satisfactory results as far as fragmentation is concerned, provided that the greatest spacing of the holes and rows in fairly hard or tough material is not more than three-quarters the depth of the cut, and that these holes penetrate sufficiently below grade to leave no hard, projecting points between adjacent holes. If the trial blast shows insufficient fragmentation the amount of explosive may be increased, or if much of the material is broken too large the spacing should be decreased. Since the burden on any hole of a given depth varies directly as the product of the spacings between holes and rows, the amount of explosive placed in each hole should be varied proportionally whenever the distance between holes and rows is increased or decreased.

Special care should always be taken to make certain that the holes along the ditch or slope lines are placed and spaced correctly and are of sufficient depth to insure full breakage to the bottom of the established ditch line. In deep cuts of difficult materials where the requirements as to slope smoothness are severe it is frequently advantageous to drill the slope or ditch holes at about the same angle as the specified side slope. Then, instead of placing the entire charge at the bottom of the hole a part is placed in the stem as a secondary charge, which is primed and wired so as to explode simultaneously with the main blast.

CLOSER SPACING NECESSARY WHERE HEAVY SHOOTING IS NOT PERMISSIBLE

Where heavy shooting is not permissible the only way in which good fragmentation can be obtained is by a closer spacing of the holes. But drilling is expensive. A foot of drill hole seldom costs less than a pound of high-grade explosive and frequently as much as 2 pounds. The spacing should therefore be the greatest at which sufficient fragmentation to permit easy shovel operation can be achieved within the limits of the heaviest permissible shooting. In very hard, tough rock this maximum permissible spacing is sometimes as low as one-half and seldom exceeds three-quarters the depth of the cut for cuts less than 16 feet deep, and probably rarely exceeds 12 feet of actual distance for cuts over 16 feet, unless supplementary charges are placed in the stem of the original or master holes and also in holes which have been drilled to within one-third to one-half the depth of the cut near the middle of each quadrilateral formed by four deep holes.

SUMMARY OF FACTS BROUGHT OUT BY STUDY

The sole purpose of drilling and blasting in highway grading work is to condition the material for removal, usually with a power shovel. Moreover, the daily or hourly cost of the power-shovel grading outfit is high—usually between $15 and $25 an hour—and practically constant whether the rate of production is high or low. Low production cost can therefore only be obtained through high production; but in order to maintain a high production rate the blasting must be thorough and

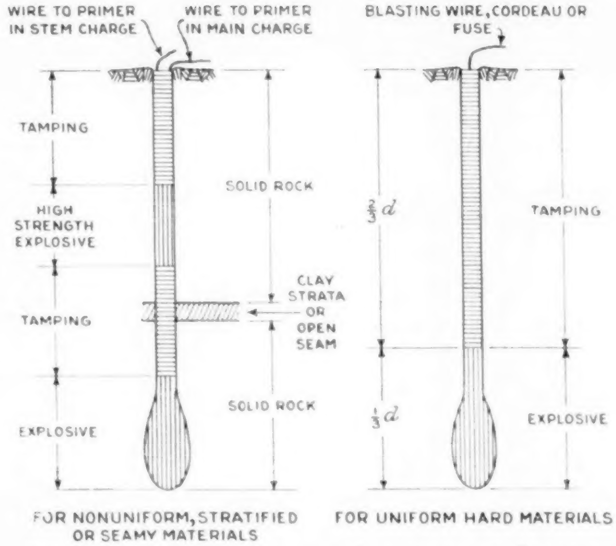

FIGURE 11.—SUGGESTIONS FOR LOCATING THE BLASTING CHARGE. HOLES SHOULD BE SPRUNG UNTIL THE DESIRED AMOUNT OF EXPLOSIVE WILL NOT OCCUPY MORE THAN ONE-THIRD OF TOTAL DEPTH. IN MATERIAL WHICH IS HARD TO SHATTER PROPERLY, A SMALL CHARGE OF HIGH-STRENGTH EXPLOSIVE SHOULD ALSO BE PLACED IN THE STEM WHENEVER THE DISTANCE TO THE NEAREST HOLE EXCEEDS 8 FEET. NO STEM CHARGE SHOULD HAVE LESS THAN 2 FEET OF TAMPING OVER IT

sufficient to shatter the material into fragments small enough for ready handling by the shovel. The blasting must also proceed sufficiently ahead of the shovel so that there will be no appreciable delay from waiting for the blasting to be done. This means, in general, that the blasting of each cut should be completed before work is begun on that cut by the shovel. On this account, road work seldom affords much opportunity for firing the various rounds against a free face as in quarry work. As a result, relatively more explosive is required and relatively closer spacing of holes is necessary in order to insure proper fragmentation.

Perhaps the most significant feature disclosed by this study of drilling and blasting in highway grading work is the very general absence of that grade of supervision, organization, and matured planning which is given to the remainder of the job. Frequently all decision as to such important points as the location, spacing, and depth of holes is left entirely to the drillers. Too often this results in the holes being placed where it is convenient to drill rather than where they will do the most good. This lack of planned direction usually also extends to the method of loading and the amount of explosives used in each individual hole. With such conditions forming the general rather than the exceptional practice, there is no wonder that nearly three-quarters of all rock jobs reported the shooting as unsatisfactory from the viewpoint of fast, easy shovel operation. All the available data indicate that the cost of this poor shooting is as high and (if we include the usual pop holes to attain grade and the block holes and mudcaps necessary to break up boulders) frequently much higher than on those jobs where the shooting is good. For similar conditions and materials, the poorly-shot jobs often actually pay more per cubic yard for their drilling and blasting than do those jobs on which

(Continued on p. 308)

EFFECT OF TYPE OF BREAKING MACHINE ON THE MODULUS OF RUPTURE OF 6 BY 6 INCH CONCRETE BEAMS

By O. K. NORMANN, Junior Highway Engineer, Division of Management, U. S. Bureau of Public Roads

CONCRETE pavements are now very generally opened when the breaking strength or modulus of rupture of beams made at the time the slab was placed reaches some specified value. Various types of portable or semiportable machine are used for breaking these beams, and the question of the comparative values given by different types of machine under actual field conditions is therefore of prime importance.

During the course of some rather extensive production studies during the summer of 1930, an opportunity arose to compare the values obtained under field conditions from two radically different types of beam-breaking machine, and to check these results against the values given by 2-point loading in a universal testing machine. These data indicate that it would be well if every field machine could be calibrated against the standard laboratory set-up before it is placed in service. As these field machines frequently receive very rough treatment it might also be desirable to have this calibration rechecked before the beginning of each subsequent construction season.

The chief structural difference in the various types of machines now in use for breaking concrete beam test specimens is in the arrangement of the lever arms, which produce the bending moment in the beam. All of these various arrangements are apparently satisfactory so far as breaking the test specimens is concerned, but differ widely in the amount of shearing stress imparted while developing the necessary bending moment.

TESTS MADE WITH THREE TYPES OF BREAKING MACHINE

During the progress of certain production studies on Tennessee Federal-aid project 212-A, in McNairy County, involving the construction of 12.34 miles of concrete pavement, it was noted that the modulus of rupture as obtained for beams broken by a portable multiple-lever beam-breaking machine was regularly much higher than values given by beams broken with a very simple cantilever device. Since two multiple-lever machines, one simple cantilever device, and a universal testing machine were available, a series of eight sets of beams were at once made and cured under conditions as nearly alike as possible in every respect. Some breaks of each set were made by the simple cantilever device; others by one or the other of the two portable multiple-lever machines, and still others were broken by the 2-point loading method in the universal testing machine.

Three 6 by 6 by 36 inch beams for each of 7 sets and one set of 4 beams, or a total of 25 beams, were made from concrete taken from near the center of selected batches of uniform consistency as they were dumped on the subgrade. The molds, made of angle irons with an iron-plate bottom, were each filled, spaded once around the sides, tamped 50 times 1 inch from the side with a standard tamping rod, and then spaded once more, after which the top surface was finished with a trowel. After being marked and numbered, the beams were covered with wet burlap and allowed to remain undisturbed in the molds until they had been hauled to the curing tank on the following day, when the molds were removed and the beams placed under water.

Beam 1 of each set was kept submerged until the twenty-eighth day, when it was removed from the curing tank and broken (three breaks) on the simple cantilever device. Beams numbered 2 and 3 of each set were removed from the curing tank after 23 to 26 days of curing, wrapped in wet burlap, and hauled by truck to the Tennessee State highway laboratories, where they were placed in the moist room until the twenty-eighth day, at which time each beam was broken once in the 50,000-pound universal testing machine and then twice in the multiple-lever type of portable beam-breaking machine. The top, or troweled surface, of the beam was always placed so as to be on the tension side.

CANTILEVER APPARATUS DESCRIBED

The cantilever apparatus used in these tests, as well as in breaking a large number of other similar beams, consisted essentially of (1) a strong upright post provided with a heavy bearing plate and a simple clamp for holding the beam firmly in a horizontal position while the bending moment was being applied; and (2) a 6-foot lever arm, one end provided with an adjustable socket for attachment to the beam and the other with a hook for suspending a 50-gallon tank to which water was added at a uniform rate until the breaking point of the beam was reached. The exact arrangement of the apparatus as used on this work is shown in Figure 1. Figure 2 shows the details of a similar apparatus which includes such modifications as three months of experience in making over 500 breaks of 8 by 6 inch beams with the first machine suggested as desirable.

The improvised machine used on this job to break the beams in the field was of no standard design or make. The advantages of this type of machine are:

(1) Simplicity of design, portability, and cheapness.

(2) The fact that the load may be applied at a uniform rate, or at any desired rate.

(3) The fact that the beam arm hangs absolutely free and is not dependent upon any ball or roller bearings. There is no system of levers that may vary in length or position. The only variable is the total weight of the water. This may be weighed to the nearest pound.

The method of operation was very simple. The concrete beam was clamped to a bearing plate on top of the wooden post by means of two ½ by 3 inch steel plates. The large bearing plate could be adjusted by means of the bolts holding it, so as to be level. The rear clamp was not tightened very hard. If it had been, there would have been a tendency for the beam not to be in contact with the bearing plate directly under the plane of maximum stress. For this reason, both of the clamps were left slightly loose in order that the beam would always be in direct contact with the breaking edge. The use of a heavy but rather narrow angle iron dapped into the front top edge of the post is suggested for future machines of this kind to replace the large bearing plate, while the two top clamp plates should be replaced by one similar angle iron. This modification is shown in Figure 2.

The beam arm was held in place by two ½ by 3 inch plates, one on the top and one on the bottom of

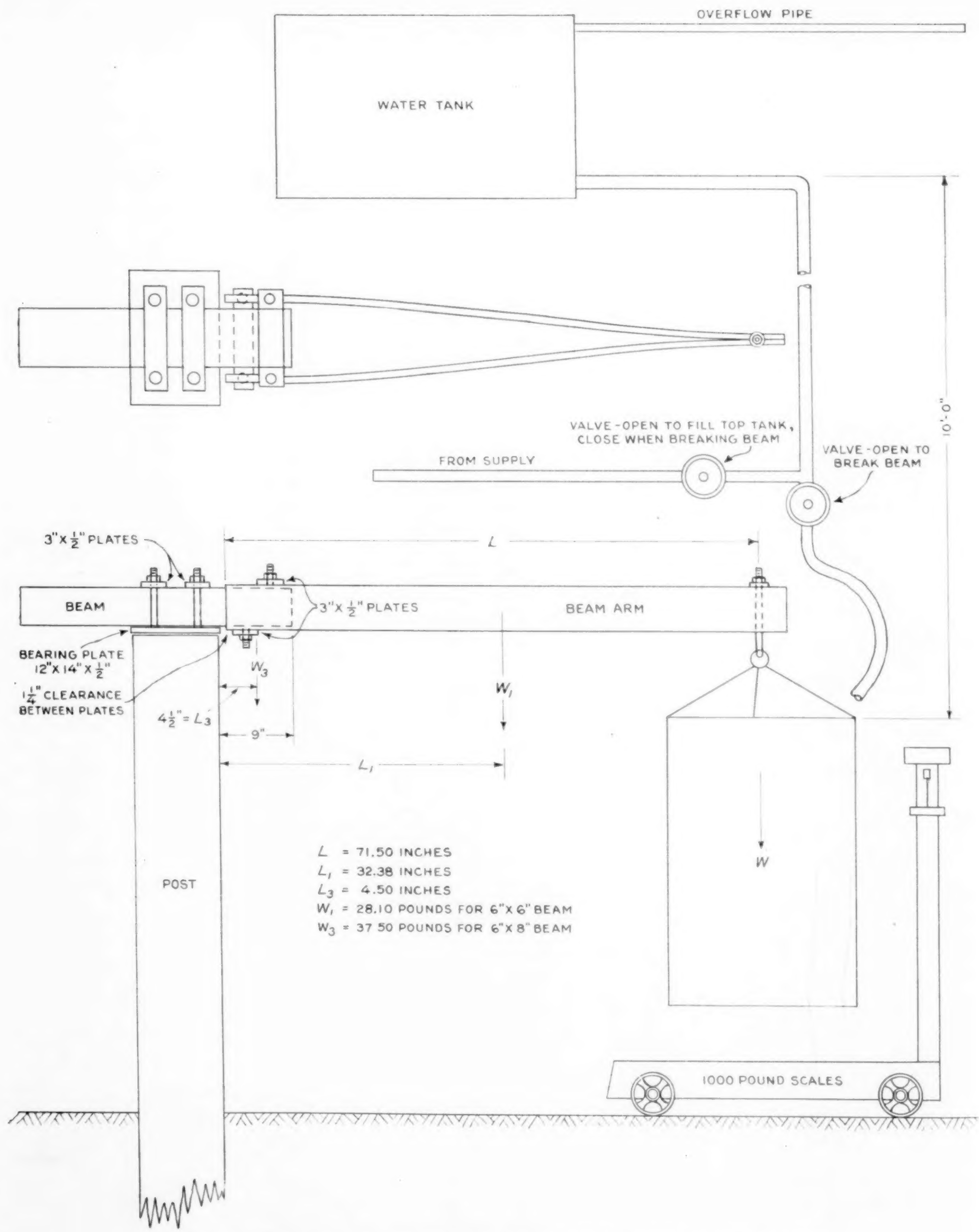

FIGURE 1.—DIAGRAM OF IMPROVISED SINGLE-LEVER FIELD MACHINE AS USED IN TESTS

the arm. The plates could be adjusted so that the beam arm would be level at the time of the break. At first the total deflection of the end of the beam arm was about 5 inches when the break took place. This deflection was later reduced to about 3 inches by placing angle irons at the end of the wooden arm, thereby reducing the play in the two plates.

Water was supplied at a uniform rate from a tank above the beam machine. This tank was filled to the same level before making each break. There was a very slight difference in the rate of application of the load at the beginning and end because of the fact that the head was reduced slightly as water was drawn from the top tank. An inflow equal to the outflow would insure a more constant rate of load application and would also permit the use of a smaller tank.

A wooden platform was placed between the scales and the tank so that the heavy tank of water would not fall directly on the scales. The tank of water could then be easily lowered on to the scales and weighed to the nearest pound. If scales are not readily available, the tank can be calibrated and the quantity measured within an accuracy of about 1 per cent. In this case a mound of sand or earth should be made so that the drop of the tank will not exceed 1 inch when the beam snaps.

Following is the method used in computing the modulus of rupture for tests with the single-lever breaking device:

In Figure 1

W = Load applied by tank.
L = Corresponding moment arm.
W_1 = Weight of beam arm and auxiliary parts.
L_1 = Corresponding moment arm.
W_3 = Weight of cantilevered portion of test beam.
L_3 = Corresponding moment arm.

Then, if M is the moment acting at the support,

$$M = WL + W_1L_1 + W_3L_3,$$

and the modulus of rupture is given by the equation,

$$S = \frac{Mc}{I} = \frac{6M}{bd^2}.$$

LOAD APPLIED AT THIRD POINTS ON UNIVERSAL TESTING MACHINE

A frame had been made by the Tennessee highway department to support a 6 by 6 inch beam, as shown in the diagram in Figure 3, so that the 50,000-pound universal testing machine could be used to break beams. The beam was supported at each end on 2-inch iron plates, and the load applied at two points on the beam by the testing machine. Cylindrical rollers were placed between the plates in contact with the beam, as well as between the plates connected with the frame through which the load was applied.

The apparatus for breaking the beams was built so that the distances between the bearing points could be changed if desired. For this study the outside supports were placed 30 inches apart, with the load applied at the two one-third points. The bending moment due to applied load was then the same for all points between the two places where the load was applied, thus providing a favorable condition for the break to occur at the weakest place within the middle 10 inches of the beam. Furthermore, while the bending moment was maximum and uniform within this section of the beam, the shearing stress due to applied load was equal to zero over the full length of the same distance. The troweled surface was placed on the bottom so that it would be on the tension side of the beam when it was broken.

Values of the modulus of rupture for tests made in the universal machine were computed by the following formula:

$$S = \frac{3WA}{bd^2} + \frac{3W_1L}{4bd^2},$$

where W is the total applied load, A the distance from each support to the adjacent loading point, W_1 the weight of the beam, and L the distance between supports. Since in these tests the loading was at the third points, the formula was reduced to the form

$$S = \frac{L\ (W + \tfrac{3}{4}W_1)}{bd^2}.$$

MULTIPLE-LEVER TESTING MACHINES DESCRIBED

Two of these machines were available, one in the laboratory and one in the field. All of the beams made for this particular study to be broken on these machines were broken in the laboratory and with the laboratory machine, except two, which were broken on the field machine. It was not intended to break any on the field machine but the results obtained on the other machines made it appear worth while to carry the study further, because there was considerable difference in the amount of shear produced in the two machines.

In both of the multiple-lever machines the load was applied by a cable attached to a hand wheel. A dynamometer between the cable and the end of the beam arm indicated the load being applied. This dynamometer was carefully tested from time to time, and was always correct within less than 1 per cent.

The chief difference between the laboratory and the field machine is the distance from the point where the load was applied to the plane of maximum bending stress. In the laboratory machine this distance is 5.5 inches, while in the field machine it is 8.75 inches. This difference causes a greater shearing stress in the one case for the same moment at the plane of maximum stress. The essential differences of the two machines are shown in Figure 4.

Following is the method used in computing the modulus of rupture for tests with the multiple-lever machines.

In Figure 4,

W_1 = Applied load.
L_1 = Distance from the upper fulcrum to the point of load application.
W_2 = Weight of beam arm.
L_2 = Corresponding moment arm with respect to upper fulcrum.
W_3 = Weight of cantilevered portion of test beam.
L_3 = Distance between upper and lower fulcrums.
W_4 = Weight of dynamometer.
L_4 = Moment arm of lower fulcrum with respect to front edge of supporting post.
L_5 = Length of cantilevered section of test beam.

The reaction at the lower fulcrum is given by the equation,

$$R = \frac{W_1L_1}{L_3} + \frac{W_2L_2}{L_3} + \frac{W_4L_1}{L_3}$$

The moment at the support is then given by the equation,

$$M = \frac{W_1L_1L_4}{L_3} + \frac{W_2L_2L_4}{L_3} + \frac{W_4L_1L_4}{L_3} + \frac{W_3L_5}{2}$$

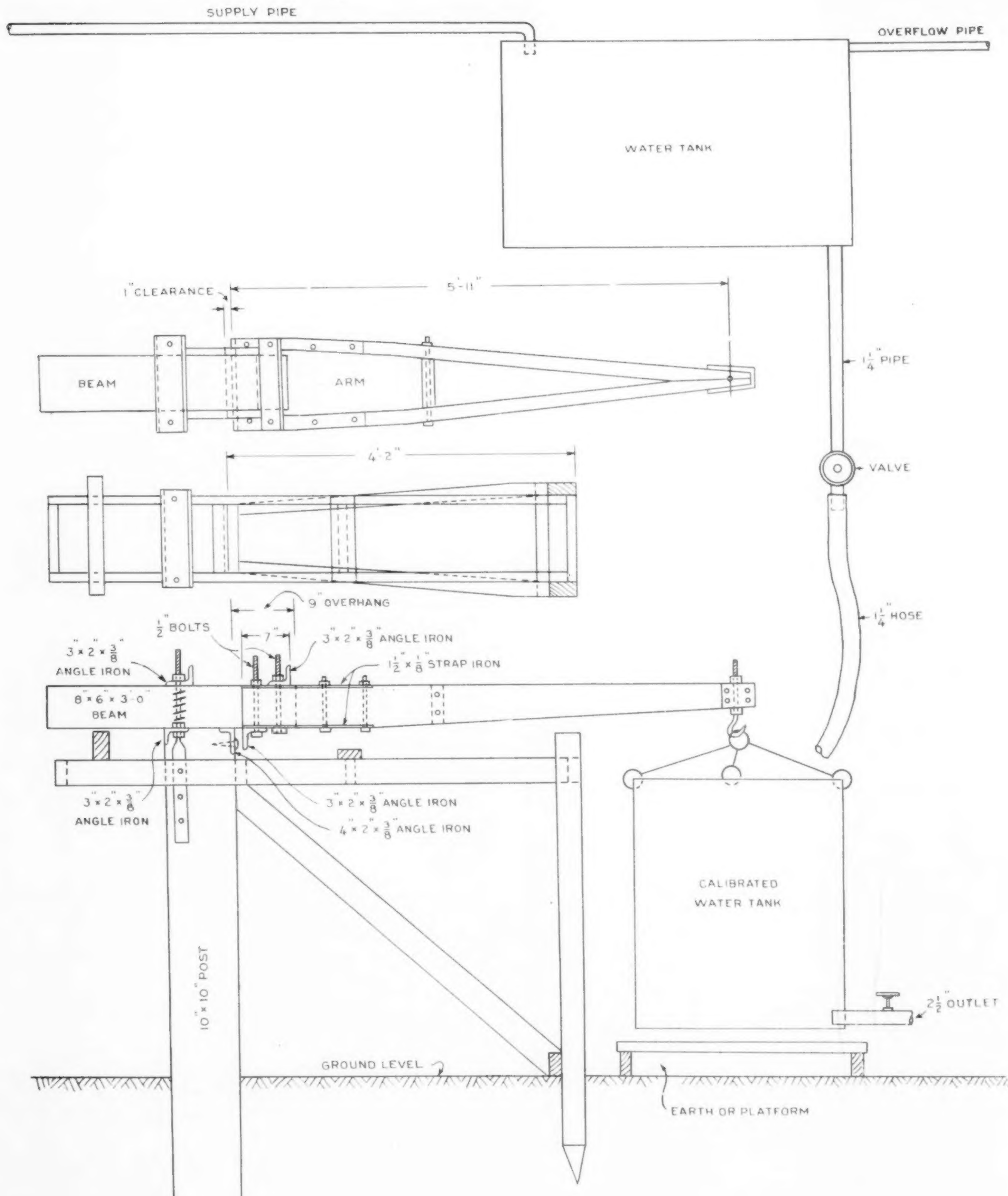

Figure 2.—Diagram of Improved Single-Lever Field Machine

and the modulus of rupture is obtained from the relation

$$S=\frac{6M}{bd^2}$$

VALUES OF MODULUS SHOW SIGNIFICANT DIFFERENCES

Values of the modulus of rupture of all beams tested in the three types of machine are given in Table 1. Table 2 gives the differences between average values for each set tested on the universal machine and the corresponding average values for tests with the single-lever and multiple-lever machines.

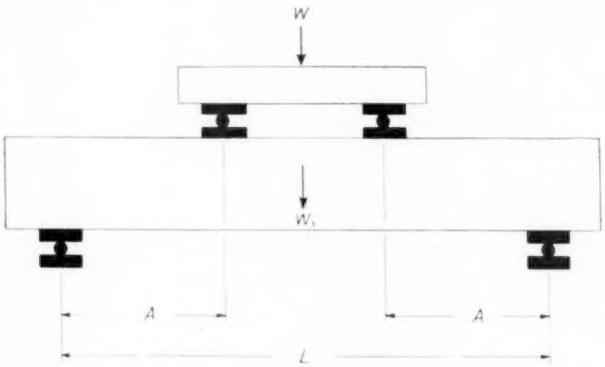

FIGURE 3.—DIAGRAM SHOWING 2-POINT LOADING OF BEAM IN UNIVERSAL TESTING MACHINE

TABLE 1.—*Comparison of values of modulus of rupture given by the three types of beam-breaking machine; modulus of rupture given in pounds per square inch*

Set No.	Improvised single-lever machine: Beam 1, break No.— 1	2	3	Average	Universal machine: Beam 2, Break No.— 1	Beam 3, Break No.— 1	Average	Multiple-lever laboratory machine: Beam 2, break No.— 1	2	3	Beam 3, break No.— 1	2	Average
5	716	614	730	687	712	666	689	865	823	860	926	841	863
6	553	563	568	561	525	559	542	759	764	----	883	780	797
9	572	575	540	562	570	611	590	683	672	----	761	868	746
12	574	586	536	565	614	576	595	756	772	773	786	781	773
14	560	517	519	532	605	546	575	856	816	----	710	786	792
16	528	654	613	598	572	565	568	788	732	----	812	867	800
20	628	633	590	617	644	569	607	950	913	----	938	899	925
Average for machine	----	----	----	589	----	----	595	----	----	----	----	----	814

TABLE 2.—*Differences in average values of modulus of rupture given by tests with universal testing machine and corresponding values given by tests with the improvised single-lever machine and the multiple-lever laboratory machine*

Set No.	Improvised single-lever machine	Multiple-lever laboratory machine
5	−2	+174
6	+19	+255
9	−28	+156
12	−30	+178
14	−43	+217
16	+30	+232
20	+10	+318
Average	23	219

The results obtained from each machine were very consistent for both the individual and the average breaking strengths. The real differences lie between the values given by the different types of breaking machine. The improvised field machine gave results very close to those given by the universal machine, the difference between the averages for all the breaks being only 6 pounds per square inch. For these two machines the maximum difference between the average strengths of any set of beams, as shown in Table 2, was only 43 pounds, with an average difference of only 23 pounds. The maximum and minimum strengths on the improvised field machine were also very close to the maximum and minimum strengths on the universal.

The laboratory multiple-beam machine gave much higher results than either the universal or the improvised field machine. The increase in average strength was 219 pounds, or 36.8 per cent over the average strength given by the universal testing machine. On the individual sets of beams the average strength was from 156 to 318 pounds per square inch higher. There was not a break made on the multiple-beam machine that was as low as the highest breaks obtained for corresponding sets on either of the other machines. The smallest and the largest difference between the lowest break on the multiple-beam machine and the highest break on the other two, for beams of the same set, were 61 pounds and 255 pounds, respectively.

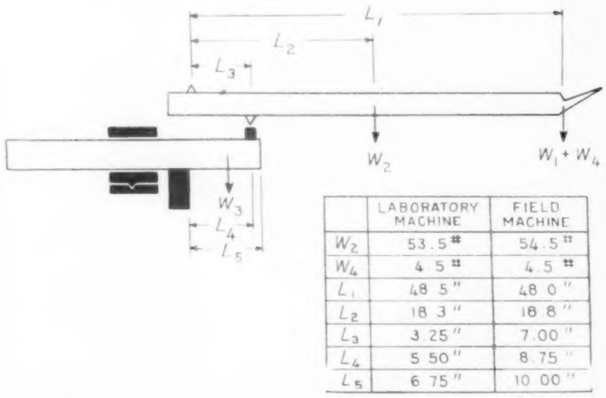

FIGURE 4.—DIAGRAM SHOWING LOADS AND MOMENT ARMS ACTING IN MULTIPLE-LEVER MACHINES

The results of the study showed that the multiple-beam machine gave results so much larger than those obtained on the other two machines that it was decided to carry the study to the multiple-beam field machine. The essential difference between the multiple-beam laboratory machine and the multiple-beam field machine is the distance from the plane of maximum stress to the point at which the load is applied to the test beam. In the first case this distance is 5.5 inches, and in the second it is 8.75 inches.

Results of tests of two beams each from set 19 on the improvised field machine and the multiple-beam field machine are given in Table 3.

All of these beams were made from the same batch. Beams 1 and 4 were tamped 50 times, and beams 2 and 3 were tamped 180 times.

The excess in average strength given by the multiple-beam field machine was 152 pounds, or 26 per cent higher than the average strengths obtained on the improvised field machine.

TABLE 3.—*Comparison of values of modulus of rupture given by tests with the multiple-lever field machine and the improvised single-lever machine, for two beams each from set 19*

Beam	Multiple-lever field machine			Improvised single lever-machine			
	Break number		Average	Break number			Average
	1	2		1	2	3	
1				550	552		551
2				639	577	617	611
3	753	770	762				
4	693	715	704				
Average for machine	733			581			

CONCLUSIONS SUMMARIZED

In view of these facts the following observations and conclusions seem justified:

1. The breaks on the beams broken in the multiple-lever machines showed that a much larger percentage of the rock was broken than in the other machines.
2. The type of machine used to make the break will determine to a large extent what breaking strength will be obtained.
3. The results show that the apparent quality of concrete, as indicated by test specimens, may be varied be merely changing the apparatus for breaking specimens. They may also help to emphasize why a more standard method of making and especially of testing beams in the field should be adopted.
4. It would be advantageous if every field machine could be calibrated against the standard laboratory set-up before it is placed in service. As these field machines frequently receive very rough treatment it might also be desirable to have this calibration rechecked before the beginning of each subsequent construction season.

Acknowledgement is made to the Department of Highways and Public Works of the State of Tennessee, without whose valuable assistance and active cooperation in both laboratory and field these data could not have been obtained.

(Continued from p. 302)

the shooting is good, and in addition carry the penalty of a low rate of production on the part of the shovel. The conclusion therefore seems inescapable that, under present close margins of bidding, the grading contractor who wishes to remain in business can not afford to permit poor shooting on his rock jobs.

ROAD PUBLICATIONS of the BUREAU OF PUBLIC ROADS

ANNUAL REPORTS

Report of the Chief of the Bureau of Public Roads, 1924.
Report of the Chief of the Bureau of Public Roads, 1925.
Report of the Chief of the Bureau of Public Roads, 1927.
Report of the Chief of the Bureau of Public Roads, 1928.
Report of the Chief of the Bureau of Public Roads, 1929.

DEPARTMENT BULLETINS

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Report of a Survey of Transportation on the State Highways of Vermont. (1927.)

Report of a Survey of Transportation on the State Highways of New Hampshire. (1927.)

Report of a Plan of Highway Improvement in the Regional Area of Cleveland, Ohio. (1928.)

Report of a Survey of Transportation on the State Highways of Pennsylvania. (1928.)

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UNITED STATES DEPARTMENT OF AGRICULTURE

BUREAU OF PUBLIC ROADS

CURRENT STATUS OF FEDERAL-AID ROAD CONSTRUCTION

AS OF

JANUARY 31, 1932

STATE	COMPLETED MILEAGE	UNDER CONSTRUCTION					APPROVED FOR CONSTRUCTION					BALANCE OF FEDERAL-AID FUNDS AVAILABLE FOR NEW PROJECTS	STATE
		Estimated total cost	Federal-aid allotted	MILEAGE			Estimated total cost	Federal-aid allotted	MILEAGE				
				Initial	Stage[1]	Total			Initial	Stage[1]	Total		
Alabama	2,405.4	$ 1,541,469.27	$ 748,838.58	62.6		62.6	$	$				$ 5,389,119.31	Alabama
Arizona	1,113.5	3,100,414.28	2,038,617.71	151.7	84.6	236.3	214,260.26	161,316.54	18.7		18.7	1,491,162.58	Arizona
Arkansas	1,919.7	3,795,361.89	1,772,437.89	86.0	72.7	158.7	362,032.53	181,016.26	17.3		17.3	1,991,752.04	Arkansas
California	2,214.2	10,488,440.29	4,635,262.51	222.0	44.8	266.8	2,245,227.41	1,086,831.59	47.1	14.1	61.2	1,804,409.96	California
Colorado	1,516.8	5,212,372.26	2,662,850.29	221.9	66.0	287.9	547,326.44	269,751.14	16.3		16.3	2,067,570.68	Colorado
Connecticut	288.5	3,084,244.77	1,181,757.90	22.9		22.9						713,252.14	Connecticut
Delaware	361.6	154,414.00	77,207.00	4.3		4.3	465,212.50	232,606.25	25.6		25.6	338,081.68	Delaware
Florida	581.7	5,239,072.77	2,432,196.81	147.0		147.0	49,004.56	24,502.28	.1		.1	3,021,969.03	Florida
Georgia	2,999.0	6,988,646.29	3,250,027.65	184.6	153.5	338.1	988,337.56	494,168.76	38.0	17.2	55.2	1,744,179.93	Georgia
Idaho	1,485.3	1,726,513.35	998,918.80	85.9	80.3	166.2	252,882.85	152,007.29	18.0		18.0	1,423,480.58	Idaho
Illinois	2,627.1	22,713,061.29	10,563,419.39	665.8	28.7	694.5	3,796,860.76	1,642,351.35	140.2	9.0	149.2	2,905,293.87	Illinois
Indiana	1,787.9	7,155,748.04	3,539,673.88	189.2		189.2	3,897,200.83	1,789,777.42	141.4	2.9	144.3	1,584,606.30	Indiana
Iowa	3,379.0	331,333.50	138,397.76		11.9	11.9	1,743,306.44	826,688.01	61.3	18.0	79.3	1,973,117.19	Iowa
Kansas	3,649.7	3,006,615.48	1,391,823.86	161.2	15.3	176.5	2,923,789.57	1,427,002.36	188.8	5.4	194.2	1,760,691.19	Kansas
Kentucky	1,871.7	2,348,515.36	1,067,373.06	137.2	4.0	141.2	577,471.72	269,284.62	13.3	41.6	54.9	1,727,826.78	Kentucky
Louisiana	1,541.7	7,216,064.49	2,803,668.96	112.6	10.6	123.2	1,295,507.93	68,270.07	.5		.5	1,866,879.52	Louisiana
Maine	706.0	1,804,346.15	789,080.76	31.4		31.4	37,546.34	14,086.00	.9		.9	1,318,971.43	Maine
Maryland	770.1	441,451.15	149,608.89	13.4		13.4	250,874.75	125,437.37	11.8		11.8	785,651.09	Maryland
Massachusetts	803.2	7,747,403.42	2,949,352.76	70.1		70.1	784,690.49	348,347.37	8.6	.2	8.8	687,103.02	Massachusetts
Michigan	1,974.3	10,437,694.69	4,523,057.69	374.3	58.2	432.5	800,781.24	378,823.37	34.0	8.5	42.5	3,168,443.32	Michigan
Minnesota	4,291.3	2,464,947.82	876,786.45	41.1	58.7	99.8	7,747,946.74	2,474,986.76	157.5	201.9	359.4	150,028.10	Minnesota
Mississippi	1,792.4	4,014,998.22	1,968,703.64	169.4	78.0	247.4	103,524.13	51,762.05	9.5		9.5	5,383,500.51	Mississippi
Missouri	2,951.2	2,333,230.66	975,887.91	62.8	3.4	66.2	1,560,466.24	703,143.85	60.1	3.5	63.6	3,047,358.46	Missouri
Montana	2,613.9	5,417,980.89	3,034,629.24	442.7	22.8	465.5	403,394.86	224,368.61	61.4	.5	61.9	3,253,079.46	Montana
Nebraska	4,152.9	4,920,915.22	2,406,758.29	206.5	76.6	283.1	84,279.58	41,975.64	2.1		2.1	2,647,063.39	Nebraska
Nevada	1,345.8	974,327.23	744,741.76	30.4	89.8	120.2						1,558,863.00	Nevada
New Hampshire	417.6	676,512.12	265,632.44	13.4	2.5	15.9						547,810.24	New Hampshire
New Jersey	582.1	5,708,263.52	1,976,522.53	59.5	.6	60.1						1,604,603.13	New Jersey
New Mexico	2,177.7	2,748,137.56	1,788,348.90	135.7	13.8	149.5	481,698.71	303,425.51	14.2	10.4	24.6	1,260,679.38	New Mexico
New York	3,239.0	12,643,900.00	5,610,500.00	282.2		282.2	3,514,800.00	1,350,000.00	68.7		68.7	4,436,572.98	New York
North Carolina	2,221.6	1,348,388.13	663,221.40	59.9		59.9	245,906.23	122,963.10	17.4	4.0	21.4	3,858,352.40	North Carolina
North Dakota	5,118.2	1,828,691.83	918,958.57	160.4	171.9	332.3	1,327,226.48	679,129.08	149.7	259.5	409.2	2,037,725.62	North Dakota
Ohio	2,829.6	7,002,996.22	2,423,620.10	100.1	29.8	129.9	1,173,200.00	572,962.46	10.7	.4	11.1	4,086,869.51	Ohio
Oklahoma	2,194.1	5,179,780.02	2,522,789.07	137.4	83.4	220.8	498,297.51	217,779.48	10.8		10.8	2,328,366.59	Oklahoma
Oregon	1,545.6	3,287,004.12	1,835,226.98	84.0	35.7	119.7	454,179.00	237,632.02	12.8	11.1	23.9	1,896,364.68	Oregon
Pennsylvania	3,012.1	4,193,848.20	1,917,622.88	45.8		45.8						5,096,998.06	Pennsylvania
Rhode Island	257.5	811,663.61	427,281.06	18.6		18.6	81,305.62	40,652.81	2.2		2.2	329,207.16	Rhode Island
South Carolina	1,997.0	2,679,138.08	1,148,707.78	63.0	43.6	106.6						1,375,312.87	South Carolina
South Dakota	4,035.8	2,935,366.20	1,625,070.82	174.8	175.3	350.1	497,615.45	300,502.58	54.3		54.3	1,267,439.45	South Dakota
Tennessee	1,662.4	848,984.19	422,357.41	24.2	8.7	32.9	437,058.09	202,974.45	13.6	6.4	20.0	3,883,841.72	Tennessee
Texas	7,590.0	13,062,766.47	5,801,194.84	653.6	163.0	816.6	2,728,062.52	1,234,664.59	197.3	46.8	244.1	6,787,271.18	Texas
Utah	1,218.6	850,176.96	452,772.23	59.5	.4	59.9	367,710.43	272,080.68	23.1	3.3	26.4	1,294,723.37	Utah
Vermont	339.1	104,832.58	52,346.06	4.3		4.3						572,077.12	Vermont
Virginia	1,851.3	2,748,074.56	1,320,166.66	144.2	16.3	160.5	160,977.42	80,488.71	5.4		5.4	2,090,486.72	Virginia
Washington	1,195.4	1,927,757.38	947,168.56	56.9		56.9	556,682.40	222,000.00	27.5		27.5	1,779,628.99	Washington
West Virginia	884.0	1,735,061.39	761,625.34	44.6	10.5	55.1	187,685.92	86,032.31	.7		.7	1,220,976.79	West Virginia
Wisconsin	2,664.8	3,119,586.51	1,166,563.12	138.5	14.3	152.8	90,647.32	45,000.00		3.5	3.5	2,681,895.81	Wisconsin
Wyoming	2,022.1	2,522,816.68	1,492,230.46	173.6	88.9	262.5	384,014.48	246,606.78		47.4	47.4	785,801.14	Wyoming
Hawaii	63.1	974,113.04	445,557.39	30.0		30.0	82,324.99	22,869.00	1.5		1.5	2,019,071.60	Hawaii
TOTALS	100,262.6	203,386,392.15	93,715,561.81	6,559.2	1,818.6	8,377.8	44,401,309.30	19,228,256.49	1,682.4	715.6	2,398.0	106,844,531.07	TOTALS

[1] The term stage construction refers to additional work done on projects previously improved with Federal aid. In general, such additional work consists of the construction of a surface of higher type than was provided in the initial improvement.

U. S. GOVERNMENT PRINTING OFFICE: 1932